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MILLING METHODS AND COSTS AT THE SAN GUILLERMO CONCENTRATOR OF THE COMPANIA INDUSTRIAL "EL POTOSI", S.A., SAN GUILLERMO, CHIHUAHUA, MEXICO

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MILLING METHODS AND COSTS AT THE SAN GUILLERMO CONCENTRATOR OF THE COMPANIA INDUSTRIAL "EL POTOSI", S.A., SAN GUILLERMO, CHIHUAHUA, MEXICO[1]

By C. A. Mehring,[2] F. J. Bacon,[3] and Oba Wiser[4]

INTRODUCTION

This paper, which describes the milling practice at the San Guillermo concentrator of the Compania Industrial "El Potosi", S.A., located at Chihuahua, Chihuahua, Mexico, is one of a series of similar papers being prepared by the United States Bureau of Mines.

The concentrator treats lead-zinc-silver ore at the rate of 2,300-2,400 tons of dry material daily and produces lead concentrate and zinc concentrate by selective flotation methods. The lead concentrate is smelted at the Chihuahua plant of the American Smelting & Refining Co., and the zinc concentrate is shipped to various European smelters.

ACKNOWLEDGMENTS

The authors acknowledge the courtesy of the management of the Compania Industrial "El Potosi" in permitting the publication of this paper; and thanks are due to H. H. Sharp, assistant general manager of the company, for valuable assistance in the preparation of the manuscript.

LOCATION

The concentrator is located at San Guillermo, 12 miles east of the City of Chihuahua, at an elevation of 5,200 feet. The mine is 6 miles southeast of the mill and is connected with the latter by a 30-inch gage, fully electrified railroad. Over a similar railroad lead concentrates are delivered

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2/ One of the consulting engineers, U. S. Bureau of Mines, and mill superintendent, Compania Industrial "El Potosi," S.A.

3/ One of the consulting engineers, U. S. Bureau of Mines, and metallurgist, Compania Industrial "El Potosi," S.A.

4/ One of the consulting engineers, U. S. Bureau of Mines, and consulting metallurgist, Howe Sound Co.

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directly to the smelter, and zinc concentrates to the Hacienda Robinson where terminal facilities are available for transfer to the National Lines of Mexico.

POWER

Power is purchased from the Cia. Agricola y de Fuerza Electrica del Rio Conchos, S.A. This company, a subsidiary of the Electric Bond & Share Co., operates a large hydroelectric plant at Boquilla, Chihuahua, and in case of water shortage is able to supplement its power from a steam plant near Torreon, Coahuila, which is owned by allied interests.

Power is transmitted from point of generation to Avalos, Chihuahua, about 6 miles west of the concentrator, at 110,000 volts, and is there stepped down to 22,000 volts for transmission to the various centers of consumption. At San Guillermo it is further stepped down to 2,300 volts, 450 volts, and 115 volts for use in the different departments.

WATER

No water is available naturally at the mill site, nor can any be obtained from the mines, which are practically dry. The nearest available water is obtained from near the Hacienda Robinson along the margin of the Chuviscar River from a number of drilled wells, 10 inches in diameter, in which water is encountered at a depth of approximately 300 feet; and from shallow wells. The former are equipped with air lifts and are used for providing drinking water and to supplement the others during dry seasons when little if any water is available in the strata lower than the river bed. The latter wells are equipped with deep-well pumps.

The water for concentrating purposes is elevated to the mill by two 5-1/2 by 12 inch Deane pumps through an 8-inch pipe line against a total head of 450 pounds.

Potable water is pumped to the mill and mine through a 4-inch pipe line by a 4 by 12 inch Worthington pump against a maximum total head of 1,150 pounds.

The water for milling purposes is slightly alkaline, having a pH value of 7.4 to 7.8, and total hardness of 22.5 grains per gallon. It is contaminated by sewage from the City of Chihuahua, the extent of the contamination depending on the volume of water actually flowing in the river. The more contaminated the water the better the metallurgical work at the concentrator. A careful study of this phenomenon by members of the research department was not productive of any positive conclusions.

Water at the concentrator is carefully conserved. Tailings are impounded for settling and clear water is returned to the mill circuit. All thickener overflows and effluents from concentrate filters are returned in like manner. This practice has resulted in large economies in reagents, and the fortunate absence of soluble salts in the ores has made the continued use of the system feasible.

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ORE TREATED

The large orebodies developed in El Potosi mine are replacements in limestone. The milling sulphides occur under the carbonate ores and grade from slightly oxidized where the sulphides start, to no oxidation at all as depth is reached.

In the upper sulphide horizons there are large bodies of hard ore, relatively low in lead and zinc and normal in silver, wherein the gangue minerals, in the order of their abundance, are magnetite, silicates, pyrite, and pyrrhotite. The lead and zinc minerals in this type of ore grind reasonably free at about 5 per cent on a 65-mesh screen. About 60 per cent of the silver is associated with the lead, and about 10 per cent with the pyrite. The remainder is intimately associated with the magnetite, silicates, and pyrrhotite, and can not be ground free, regardless of the degree of grinding.

In the large orebodies of the lower horizons the magnetite is absent, silicates are less abundant, and an increase in the lead and zinc content and a large percentage increase in pyrrhotite is noted, while the pyrite remains about the same. The ore as mined for milling is contaminated to some extent by limestone. The lead and zinc minerals in this type of ore are massive and grind reasonably free at about 15 to 20 per cent on a 65-mesh screen. Eighty-five to 90 per cent of the silver is associated with the lead mineral. The pyrrhotite constitutes from 35 to 40 per cent of the total weight of the ore, and is very magnetic, being easily removed with an ordinary hand magnet. It is a very unstable compound and begins to oxidize immediately when exposed to weathering. The zinc mineral in all of the El Potosi sulphide ore is strictly a marmatite, having the approximate formula for the pure mineral of FeS.3(ZnS), or 51.6 per cent of Zn, 14.7 per cent of Fe, and 33.7 per cent of S. It is glossy black in color before grinding and changes to a chocolate brown on being ground.

As received at the mill the ore contains slightly less than 2 per cent of moisture.

The following analysis of a composite sample of mill heads for the year 1931 is typical of ore from the lower levels:

Ounces		Per cent									
Au	Ag	Pb	Zn	Fe	S	CaO	Ins	Cd	Mg	Mn	Cu
Trace	8.14	10.5	11.2	28.4	26.4	8.70	2.80	Tr	0.13	0.60	0.06

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HISTORY OF CONCENTRATOR OPERATIONS

Mining in the Santa Eulalia district, 18 miles southeast of Chihuahua, Mexico, in which the El Potosi mine is located, began in 1703.

During the many years the El Potosi Mining Co. was removing the large tonnages of oxidized lead-silver ores for direct shipment to the lead smelter of the American Smelting & Refining Co., 6 miles southeast of the City of Chihuahua, they had developed large tonnages of low-grade complex silver-lead-zinc sulphide ores under the oxidized ores and in the lower limestone horizons of the mine.

In 1920 sufficient total tonnage of sulphide ores had developed to warrant the construction of a milling plant, providing a satisfactory treatment method could be worked out. At that time metallurgists were still handicapped by lack of knowledge of flotation reagents and control in the milling of complex sulphide ores.

At the Hacienda Robinson, the headquarters of the company, a pilot mill was built in 1922 having a capacity of 50 tons per day, to work out the flow sheet on which to base the design and construction of a larger plant. This mill was equipped with crusher, rolls, various concentrating devices such as classifiers, thickeners, tables, jigs, and several types of flotation machines and was operated intermittently until June, 1924, by applying the "then known" practice to the problem, with the result that fairly satisfactory recoveries of silver, lead, and zinc were obtained, and marketable products were made that would return a profit on the whole operation of mining and milling.

A steel and concrete structure, containing equipment to handle 550 tons of ore a day, was erected at San Guillermo. The first unit of the new plant was placed in service June 4, 1925, while a second unit was placed in service July 21.

As development work progressed in the mines, large new tonnages of milling sulphide ores were opened, and the capacity of the concentrator was gradually increased to the present rate of more than 2,500 tons daily.

Mention should be made that in the year prior to the starting of the new mill metallurgists had made remarkable advances in treating the complex ores of silver, lead, and zinc by the discovery of a number of new reagents and by better application of old ones, which gave a much brighter outlook for success in the new plant when compared with the results that had been obtained in the pilot plant.

From the starting of operations in June, 1925, and up to July 31, 1932, a total of 2,528,009 dry tons of ore have been treated.

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Crushing

The coarse crushing of El Potosi ores through 3-inch ring size is done at the mine in a 36 by 24 inch Allis-Chalmers ore breaker. The product was delivered to the mill and crushed through 5/8-inch opening in a 4-foot Symons vertical disk crusher until November 1, 1928, when a 5½-foot Symons cone crusher was installed. This crusher has a capacity of slightly over 2,000 tons per day of oversize from the Hum-mer screen, which it crushes to 3/8-inch size. The Hum-mer screen, which is placed ahead of the cone crusher in the flow sheet, has openings of ½ by ¾ inch, the undersize falling directly to the fine-ore conveyor.

Fine Grinding

Grinding practice has remained substantially the same throughout the whole period of operations. The original equipment consisted of four 7 by 12 foot Cole-Bergman rod mills, one 6-foot Dorr duplex classifier being in circuit with two rod mills. In 1927 two 5 by 12 foot Marcy rod mills, each in circuit with a 6-foot Dorr duplex classifier, were installed to assist in handling the increasing tonnages.

Lead Flotation Practice

The unthickened overflows from all Dorr classifiers have constituted the feed to lead flotation. Minerals Separation Sub-A type machines were used at the start for floating the lead. Later installations to handle the increasing tonnages were the Standard Minerals Separation spitz type of machine with the El Potosi modification. Due to the high specific gravity of the ore, the standard design of Minerals Separation machine would not circulate the pulp without choking. To meet this condition the regular feed pipes which advance the pulp from cell to cell were eliminated and short elbows were substituted, connecting the lower part of the spitz with the bottom of the agitation chamber of the same cell, thus circulating the feed within the cell. Pulp was advanced through the machine by a hole cut through the partition of each cell, at a height to maintain the proper pulp level in the spitz.

Feed to lead machines contains about 45 per cent solids.

No real cleaning of the El Potosi lead concentrates has ever been practiced, as the character of the ore, the rod mill grinding (producing a minimum of slime), and the adjustment of oils and reagents have all been favorable for the production of a relatively high-grade concentrate from the single operation of roughing. Lead concentrates from the roughers are sent direct to thickeners and thence to filters.

Past practice in the use of oils and reagents involved numerous changes leading up to the present practice. At the start, sulphurous acid was selected as the most suitable depressant for zinc, as during the first three years of milling the bulk of the ore came from the upper levels in which a large percentage of the silver was associated with the pyrite, and in order

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to make a satisfactory silver recovery it was necessary to float the latter. The acid was added in water solution and ground with the ore, or to the classifier overflows, depressing both the zinc and the pyrite. However, upon addition of a sufficient quantity of xanthate, the depressing effect on the pyrite was nullified without affecting the zinc, thus permitting the flotation of the silver-bearing pyrite.

In July, 1928, tests were made which indicated that the ores had changed to such an extent that cyanide could be used to produce a higher grade of lead concentrate. For a period of over six months after the adoption of this reagent the silver recovery fell off as compared with the sulphurous acid period, but the lead concentrates were of higher grade, and thus compensated for the silver loss. With the steady increase of ores from the lower horizons of the mine, wherein the main portion of the silver was associated with the lead, the silver recovery increased correspondingly. The addition of the chemical just ahead of lead flotation produced results identical with those obtained when it was added directly to the rod mills and ground with the ore. The latter method was adopted. The use of soda-ash along with the cyanide had to be discontinued, as the return water seemed to promote the flotation of zinc in the lead circuit. The waters used for milling purposes are alkaline after boiling off the free carbon dioxide; hence, adding cyanide in small quantity to a raw water of this character gave reaction products that were highly favorable for depressing both the zinc and pyrite in ores of this type, and consequently the addition of soda-ash was not necessary.

Potassium ethyl xanthate was used initially for promoting lead flotation and was added to the pulp at the head of the lead circuit. During this period of the operation a considerable percentage of coarse lead sulphide and also what might be termed lead middlings, passed through both lead and zinc flotation and was lost in the final tailing. Upon investigation of these losses a tabling plant was installed to treat the tailing, and both lead and silver recovery were substantially increased. Shortly after this plant was placed in service, manufacturers began to produce higher alcoholic xanthates. The adoption of potassium amyl xanthate immediately gave a vertical increase of approximately 2 unit per cent in recovery of both lead and silver in addition to recovering practically all that had been recovered by the tables. Hence, the operation of the tabling plant became unprofitable.

When using ethyl xanthate, cresylic acid was used as the frother for lead and was added with the ethyl xanthate at the head of flotation. It had been added at various times in the grinding mills with no difference in results. Shortly after the change was made from ethyl to amyl xanthate, cresylic acid was again added to the grinding mills, with a resulting increase of 6 unit per cent in grade of concentrate, obtained simply by changing the point of addition. Advancing the cresylic acid to this point also gave 1 unit per cent higher recovery of lead and silver. This brings out the point that when one of the important reagents is changed for another the change may also affect the status of the other reagents.

In July, 1932, butyl xanthate was substituted for amyl xanthate, effecting a saving in reagent cost and giving equal metallurgical results.

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During the first four years of operation, lime was not used except in an experimental way. Numerous tests, made in both laboratory and mill, wherein lime was ground with the ore gave much lower recoveries of both lead and silver, although the grades of concentrates were excellent. At the time these tests were made the cresylic acid was being added at the head of lead flotation. Later, as noted previously, it was added to the grinding mills and ground with the ore. Conditions were now favorable for a reversal of the former procedure, therefore with cresylic acid in the mills the lime was added to the pump ahead of flotation, producing slightly higher recoveries of lead and silver without affecting the grade of concentrates. The final outcome indicates that lime, in moderate amounts, has no depressing effect on lead sulphides, provided they are oiled before it is added.

The quantity of lime added before lead flotation is regulated to give a hydrogen-ion value of 7.9 to 8.1 on samples of lead tailing pulp. pH determinations are made at 30-minute intervals. Close control of hydrogen-ion concentration has resulted in a very constant froth condition. A La Motte Roulette Comparator is used, usually with Cresol Red as the indicator.

Both the mat and matless types of flotation machine, using air, were given exhaustive tests on full mill scale for both lead and zinc roughing, but they failed to compared favorably with the machines now in use. Comparatively coarse grinding releases a high percentage of all economic minerals, and air machines failed to float the coarser sizes.

Oils and Reagents, Lead Circuit

A review of past milling practice on El Potosi ores shows the following oils and reagents which were used per ton of ore milled for the flotation of lead:

Reagents added at mills and ground with ore		Pounds per ton of ore
Sulphur dioxide in water solution	(July, 1925, to July 21, 1928)	0.20
Sodium cyanide	(July 21, 1928, to date)	.083
Cresylic acid	(July 26, 1930, to date)	.054
Reagents added at the head of lead flotation		
Potassium ethyl xanthate	(July, 1925, to Oct. 1, 1929)	.25
Potassium amyl xanthate	(Oct. 1, 1929, to July, 1932)	.128
Cresylic acid	(July, 1925, to July 26, 1930)	.09
Lime	(April, 1931, to date)	.712
Butyl xanthate	(July, 1932)	.125

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Zinc Flotation Practice

The tailings from the lead flotation machines, 40 to 45 per cent solids, are the feed to zinc flotation. Up to November 18, 1930, about 25 per cent of the feed was treated in the Sub-A type of machine and the remainder in the Minerals Separation Standard machine with the El Potosi modification. The latter type of machine was adopted November 18, 1930, for all zinc roughing and cleaning. Prior to this time, all lead tailing passed directly from the last lead cell into the first zinc cell, with practically no time of contact for oils and reagents before the flotation of the zinc was started. Copper sulphate and xanthate were added in the first zinc cell. Oils were used experimentally and appear to have had no favorable influence in the zinc circuit. Beginning November 18, 1930, all lead tailings were collected in a common sump before mechanical distribution to the zinc roughers. This arrangement gave about five minutes contact of reagents with the pulp ahead of flotation. The copper sulphate and ethyl xanthate were added in the sump ahead of distribution to machines. Aryl xanthate could not be used for zinc flotation, as it had a tendency to float the pyrrhotite. Beginning in May, 1929, lime was added in one of the cells near the end of the lead circuit after the main portion of the lead had been floated. Applying it at this point depressed both the pyrite and pyrrhotite slightly and thus gave a little higher grade of rougher zinc concentrate and satisfactory zinc recovery. This practice was continued to November 18, 1930. At that time it was added to the lead tailings sump before distribution to the zinc roughers, thus giving five minutes contact. The pyrite was depressed, but the pyrrhotite only slightly so, and the only way more time contact could be obtained was to add the lime at some point ahead of lead flotation, a previous practice which had lowered both lead and silver recovery. However, under prevailing conditions, the cresylic acid being added in the grinding mills, lime could be added at the sump ahead of lead flotation. As already noted under "Lead flotation practice" the addition of lime at this point was highly beneficial to lead and silver recovery and in addition to this the 35 minute time of contact of the lime with the pulp in its passage through the lead flotation depressed the pyrite and pyrrhotite almost completely, producing a high-grade zinc rougher concentrate and high rougher recovery of zinc.

Cleaning of Zinc Concentrate

Cleaning of the zinc concentrate was not undertaken until 1927. As stated previously in this report, the matless type of air flotation machine failed as a rougher because it would not float the plus 65-mesh coarse zinc, amounting to 15 per cent of the total that could be recovered with the Standard Minerals Separation machine. The matless air machine was used as a zinc cleaner for a period of two and one-half years, producing a fair grade at the expense of losing most of the coarse mineral that had been floated by the roughers. Returning the cleaner tailings to the rougher circuit built up a circulating load of coarse mineral in the cleaner. The failure of the matless air machine in all cases may have been due to the highly granular character of the several products treated. Neither cyanide nor lime were of any value in this machine. Copper sulphate in proper quantity, about

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0.30 pound per ton of material being treated, was used and functioned to depress both the pyrite and lead.

In December, 1929, a Standard machine, El Potosi modification was available for cleaning, and in June, 1930, another of the same type was available for recleaning. Both grade and recovery of zinc were greatly improved. Neither lime nor copper sulphate had any value in the cleaning circuit with these machines. The feed was slightly alkaline. Oils had no value, but cyanide, added at the head of the first cleaner to equal 0.10 pound per ton of material being treated, had good value. No reagents were used in the second cleaner. The disposition of cleaner tailings is covered in this report under "Present mill methods."

Tabling of Zinc Concentrate

Zinc concentrates were all passed over Wilfley tables until the time when cleaning was undertaken in the Standard flotation machine, the tables functioning both for pilot purposes and for removing as much lead as possible. The source and character of this lead is given under "Lead Flotation." After the adoption of amyl xanthate very little free lead sulphides escaped from the lead circuit, so tabling was unnecessary.

Oils and Reagents, Zinc Circuit

The reagents used in past practice for roughing of El Potosi zinc were as follows:

	Pounds per ton of ore
Copper sulphate (to November 18, 1930)	0.83
Ethyl xanthate (to November 18, 1930)	0.11

For reagents employed after the above date, refer to "Present milling methods."

Milling Costs and Concentrator Results

Milling costs for several years operation are given in Tables 5 and 6. Concentrator results for seven and a half years beginning with 1925 are presented in Table 6.

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Table 1.- Concentrator results, El Potosi ores, from records of previous years

Year	Dry short tons	Ag, oz. per ton	Lead, per cent	Zinc, per cent	Recovery, per cent		
					Silver	Lead	Zinc
Concentrator Heads							
1925	34,915	8.30	9.03	9.00	-	-	-
1926	118,311	8.71	10.26	9.62	-	-	-
1927	247,794	8.86	9.33	8.67	-	-	-
1928	320,287	8.67	9.07	8.29	-	-	-
1929	401,012	8.01	8.82	8.29	-	-	-
1930	469,900	8.40	10.11	9.71	-	-	-
1931	597,673	8.62	10.91	11.29	-	-	-
1932 (150 days)	337,863	8.59	11.26	11.48	-	-	-
Lead Concentrate							
1925	6,163	35.52	40.58	8.83	69.12	79.35	-
1926	20,342	37.11	51.26	5.98	75.42	85.91	-
1927	42,466	37.01	46.35	6.53	73.26	86.32	-
1928	48,463	40.12	50.93	5.08	71.77	86.67	-
1929	58,591	40.26	53.76	5.12	73.84	89.25	-
1930	74,638	44.75	60.00	5.12	84.02	93.53	-
1931	87,207	47.59	66.27	4.43	87.14	95.94	-
1932 (150 days)	53,064	48.09	67.94	3.71	87.84	95.65	-
Zinc Concentrate							
1925	4,539	4.73	4.23	36.81	-	-	53.16
1926	19,166	3.42	2.79	40.71	-	-	68.59
1927	32,625	3.99	3.07	42.33	-	-	64.26
1928	39,182	3.82	2.50	43.55	-	-	64.29
1929	47,380	2.89	1.86	45.17	-	-	64.37
1930	72,690	1.97	1.11	46.52	-	-	71.73
1931	110,983	1.97	1.01	46.44	-	-	83.09
1932 (150 days)	69,208	1.72	0.93	46.53	-	-	82.88
Tailing							
1925	24,213	2.81	1.89	3.83	-	-	-
1926	78,803	2.29	1.49	2.99	-	-	-
1927	172,703	2.69	1.27	2.76	-	-	-
1928	232,642	2.72	1.25	2.95	-	-	-
1929	295,041	2.40	1.00	2.99	-	-	-
1930	322,572	1.51	0.70	2.80	-	-	-
1931	399,483	1.21	0.43	1.90	-	-	-
1932 (150 days)	215,591	1.12	0.50	2.19	-	-	-

Note: The results as shown for the year 1932 include the first seven months of operation, all that were available at the time this report was written.

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PRESENT CONCENTRATOR METHODS

A general flow sheet of the concentrator is given in Figure 1. The method of concentration is all flotation for the selective separation of lead and zinc minerals from the original pulp.

Crushing

All primary crushing is done at the mine, and the crushed ore is delivered to concrete ore bins at the mill. It is drawn off here through one Allis-Chalmers 30-inch by 8-foot apron-type feeder from the 440-ton bin, and one Stephens-Adamson 18-inch by 7-foot, apron-type feeder from the 2,200-ton bin to a 24-inch conveyor where it is passed over a 4 by 5-foot Hum-mer screen, set at an inclination of 32°, with openings of $\frac{1}{2}$ by $\frac{3}{4}$ inch. Undersize falls directly to an 18-inch conveyor to fine-ore bins. Oversize is conveyed to a 5½-foot Symons cone crusher, driven by a 125-hp., 575-r.p.m. induction motor, direct-connected. Product of the crusher drops onto the fine-ore conveyor, where it joins the Hum-mer screen undersize, and is conveyed to either of the two fine-ore bins. These bins are cylindrical steel tanks 22 feet in diameter, one 28 feet high and the other 23 feet high, having respective capacities of 600 and 500 tons. Fine ore is drawn from the bins by means of two roller-type feeders under each unit, and is conveyed to the rod mills by means of a 12-inch conveyor to Section 1 and a 14-inch conveyor to Sections 2 and 3.

Three Merrick weightometers are used to weigh the ore, one on the conveyor carrying all coarse ore to the crusher, and one on each of the two fine-ore conveyors to the rod mills. These serve as a double check on tonnage.

Grinding

The fine-grinding equipment consists of four Cole-Bergman single-trunnion rod mills 7 feet in diameter by 12 feet long, each pair operating in closed circuit with a model "D", 6 by 22-foot Dorr classifier; and two 5 by 12-foot Marcy double-trunnion rod mills, each mill operating in closed circuit with a model D, 6 by 22-foot Dorr classifier. The Cole-Bergman mills operate at a speed of 16 r.p.m. and are belt-driven by 150-hp., 900 r.p.m., synchronous motors. Marcy mills operate at a speed of 22 r.p.m. and are driven by 125-hp., 365-r.p.m., induction motors, direct-connected through herringbone pinion and gear.

All mills are lined with manganese-steel liners and use 3-inch by 12-foot steel rods, both of Mexican manufacture. Liner consumption is approximately 0.16 pound per ton of ore, and rod consumption is about 0.88 pounds per ton of ore.

Each classifier is operated at a speed of 28 strokes per minute, Cole mill classifiers being driven by 10-hp. motors and Marcy mill classifiers by 5-hp. motors.

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Lead Flotation

The classifier overflows, which comprise the lead flotation feed, flow by gravity to a 6-inch Wilfley pump which discharges into a 4-way mechanical distributor from which the pulp flows by gravity to four flotation machines.

The equipment in the lead flotation section consists of four 14-cell, 18-inch Minerals Separation Sub-A flotation machines. Air is supplied by a Roots blower at a pressure of 2 pounds per square inch. No. 1 and No. 2 machines are driven by Tex-ropes, with one 10-hp. motor driving two spindles. No. 3 and No. 4 are driven by one 112-kw., 900 r.p.m., synchronous motor through gears and shafting. All spindles run at a speed of 315 r.p.m.

The kinds and amounts of reagents added at the rod mills and ground with the ore are:

	Pounds per ton of ore
Cresylic acid	0.056
Sodium cyanide	.050

Those added after grinding, ahead of lead flotation are:

		Pounds per ton of ore
	Lime	0.479
	Amyl xanthate	.110
or	Butyl xanthate	.110

Lime is added ahead of lead flotation in sufficient amount to give a pH value of from 7.9 to 8.1 on pulp samples taken from the tailing of lead machines, a La Motte Roulette hydrogen-ion comparator being used for making pH determinations. This degree of alkalinity tends to inhibit the flotation of iron minerals. Until recently, amyl xanthate has been used as the collector of lead and silver minerals, but at present butyl xanthate is being used for that purpose. The frothing agent is cresylic acid. Pulp density at the head of the lead circuit is maintained at about 45 per cent solids.

The character of the ore delivered to the concentrator is very uniform from day to day. It is an exceptionally clean ore and breaks in granular form with very little slime. If grinding is very fine, slime seems to cover the surfaces of the lead mineral, causing it to float with difficulty. Flotation lead concentrates can readily be made as high as 72 per cent, but the best economic grade is about 70 per cent.

Zinc Flotation

Tailing from the lead flotation machines flows by gravity to a 4-inch Wilfley pump, which elevates it to a 4-way mechanical distributor. From the distributor it flows by gravity to four 12-cell, 24-inch Minerals Separation

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Standard machines, where a rougher concentrate assaying about 40 per cent of zinc from all cells is produced. The rougher concentrate is pumped by a 4-inch Wilfley pump to the first cells of two 6-cell, 24-inch Minerals Separation machines which serve as first cleaners, where a concentrate assaying about 45 per cent of zinc is produced. This concentrate flows by gravity to two 8-cell, 24-inch Minerals Separation machines for second and final cleaning. Zinc concentrate from these machines has an average assay of about 46.5 per cent of zinc and 15.2 per cent of iron. As explained under "Ore treated," the zinc mineral is a very low-grade marmatite. Laboratory cleaning tests to determine the maximum possible grade of zinc concentrate, to the extent of seven cleanings with iron depressants, gave a concentrate assaying 59.2 per cent of zinc and 14.9 per cent of iron.

Following are the kinds and amounts of reagents used for the flotation of rougher zinc concentrate, added at the 4-inch Wilfley pump ahead of the pump distributor to flotation machines:

	Pounds per ton of ore
Copper sulphate	0.424
Ethyl xanthate	.125

Copper sulphate is added to reactivate the zinc, and ethyl xanthate is the collector. Very careful control of the copper sulphate and xanthate is necessary for satisfactory results.

Reagent used for first cleaning:

Sodium cyanide ..... 0.010 pound per ton of ore.

Dewatering and Filtering of Flotation Concentrates

Lead concentrate flows by gravity to a pump, which elevates it to a 3 by 15-foot Dorr simplex classifier mounted above the lead thickener tanks. Overflow from the classifier flows to two 8 by 30 foot steel Dorr thickeners, where it is settled and pumped by Dorrco diaphragm pumps to two 6-foot diameter, 4-lead American filters. Coarse lead or oversize from the classifier flows by gravity directly to these two filters. They handle about 335 tons of lead concentrate per day, the cake having a moisture content of about 7.5 per cent.

Zinc concentrate is pumped to a 3 by 15 foot Dorr simplex classifier operating at a speed of 22 strokes per minute, mounted above the zinc thickener. Overflow from the classifier flows to a 6 by 45-foot Dorr traction thickener where it is settled and raised to an 8 by 12-foot Dorrco filter, by means of two Dorrco diaphragm pumps. Coarse zinc concentrate, or oversize from the classifier, flows directly to the same filter. Moisture content of the Dorrco filter cake is about 7.0 per cent. This filter handles about 500 tons of zinc concentrate per day, with high peaks for a few hours at the rate of 600 tons per day.

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Thickener feed contains about 36 per cent solids, and thickener discharge about 65 per cent solids.

Cake from filters drops to 16-inch conveyors which carry it to railroad cars, or by means of a trimmer it is dropped to the patio if necessary. Cars have a capacity of 13 tons and are of the side-dump type. Lead concentrate is shipped to the American Smelting & Refining Co. smelter at Avalos; and zinc concentrate is sent to the Hacienda Robinson for transfer to cars of the National Railroad.

Filter medium is similar to American 21 twill, and is purchased from J. de la Torre y Hijos, Mexico City.

All overflows of Dorr thickeners are collected in a common sump and pumped back to the mill surge tank by means of a 4 by 3-inch Allis-Chalmers centrifugal pump, automatically controlled by float switch, having a capacity of 400 gallons per minute against an 80-foot head. This pump is direct-connected to a 10-hp. motor running at a speed of 3,300 r.p.m.

Sampling

All samples are cut at 15-minute intervals by Geary-Jennings Automatic Samplers. Eight-hour samples weigh about 15 pounds each. A portion of each sample is set aside for the monthly weighted composites. These samples are placed in air-tight containers and at the end of the month screen analyses, chemical analyses, and laboratory tests are made. Tests and analyses serve as a check against daily results.

As stated previously, the tonnage of ore milled is obtained by means of three Merrick weightometers. Daily assays are calculated into recoveries with the bi-metallic formula. Ratios of concentration give the weights of concentrates. As the samples are all taken with automatic samplers the calculated weights of concentrates and actual shipments check very closely. All concentrates leaving the mill are weighed in the cars by track scales, giving an additional check on metallurgy.

Mixing and Feeding of Reagents

All reagents except copper sulphate and lime are fed by Geary reagent feeders. Copper sulphate is fed by means of a feeder having cups attached to a revolving disk. All reagents except lime are fed as liquids. A belt-type dry feeder is used for lime.

In preparing lime for use in the mill just enough water is added to slake it, after which it is screened on 1/8-inch mesh for use in the feeder.

Copper sulphate is dissolved in two concrete mixing tanks, each having a capacity of 10,000 gallons. In order to provide for thorough mixing the solution is circulated from the bottom of the tank to the top by a small centrifugal pump made of corrosion-resisting material.

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The xanthates are made up as 10 per cent solutions in concrete tanks having a capacity of 1,280 gallons each. A 5 per cent solution of sodium cyanide is used, the solution being kept in a tank having a capacity of 2,750 gallons. All liquid reagents flow by gravity to the feeders.

Tailings Disposal

All tailings from the mill flow by gravity for a distance of 3,174 feet through a cast-iron lined launder to two 6-inch Wilfley sand pumps (one used as a spare), driven by 2,200 volt, 50-hp. induction motors, and are elevated to the face of the dam for impounding of solids and recovery of water. The discharge pipe is 6 inches in diameter, 750 feet long, with 700 feet along the outer edge of the dam. Pipe is fitted with welded nipples 18 inches in diameter by 18 inches long spaced at 4-foot intervals, and material can be deposited at any place along the face of the dam, keeping the bank or border built up sufficiently high to impound and settle out clear water. This pipe is level, and is raised 10 feet at a time over its whole length.

About 100 feet from the face of the dam a penstock with adjustable weirs skims off the clear water, which flows through an 8-inch pipe to the clear water sump for return to the mill.

Two belt-driven triplex pumps, one 4 inch and one 3 inch, return about 170 gallons per minute of recovered water through a 5-inch pipe line to the 8 by 20-foot mill surge tank for re-use. The two triplex pumps are driven by a 2,200-volt, 50-hp. motor.

Due to the high specific gravity and coarseness of the material and to rapid oxidation of the pyrrhotite the dam is solid and very little trouble is had with sliding of the border.

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Table 2.- Screen analyses of various mill products

Screen size	Coarse ore to crusher		Crusher product		Midd. feed		Rod-mill discharge		Classifier sand return		Classifier overflow	
	Weight, per cent	Cumulative weight, per cent	Weight, per cent	Cumulative weight, per cent	Weight, per cent	Cumulative weight, per cent	Weight, per cent	Cumulative weight, per cent	Weight, per cent	Cumulative weight, per cent	Weight, per cent	Cumulative weight, per cent
On 4.0-inch ring	4.00	4.00	None	None	None	None	None	None	None	None	None	None
On 3.0 do	4.43	8.43	do	do	do	do	do	do	do	do	do	do
On 2.5 do	6.23	14.66	do	do	do	do	do	do	do	do	do	do
On 2.0 do	5.39	20.05	do	do	do	do	do	do	do	do	do	do
On 1.5 do	8.78	28.83	do	do	do	do	do	do	do	do	do	do
On 1.0 do	10.51	39.84	do	do	do	do	do	do	do	do	do	do
On 0.5 do	14.38	53.72	4.44	4.44	5.2	5.2	do	do	do	do	do	do
On 4 mesh	14.08	67.80	55.21	59.65	42.3	47.5	do	do	do	do	do	do
On 8 do	8.12	75.92	11.43	71.08	15.7	63.2	do	do	do	do	do	do
On 10 do	3.45	79.37	5.10	76.18	5.5	68.7	do	do	do	do	do	do
On 14 do	3.03	82.40	4.10	80.28	5.0	73.7	do	do	do	do	do	do
On 20 do	2.51	84.91	5.83	86.11	4.4	78.1	do	do	do	do	do	do
On 28 do	2.67	87.57	2.46	88.57	4.5	82.6	do	do	do	do	do	do
On 35 do	1.77	89.35	1.37	89.94	3.1	85.7	6.4	6.4	9.6	9.6	2.5	2.5
On 48 do	2.00	91.35	1.69	91.63	2.5	88.2	10.2	16.6	18.3	27.9	5.6	8.1
On 65 do	1.90	93.25	1.60	93.23	3.1	91.3	14.4	31.0	24.2	52.1	10.2	18.3
On 100 do	1.51	94.76	1.41	94.64	2.3	93.6	15.0	46.0	16.1	68.2	12.3	30.6
On 150 do	1.13	95.89	1.08	95.72	1.4	95.0	9.6	55.6	9.7	77.9	10.8	41.4
On 200 do	1.06	96.95	1.02	96.74	1.4	96.4	13.8	69.4	8.0	85.9	10.6	52.0
Through 200 do	3.05	100.00	3.26	100.00	3.6	100.0	30.6	100.0	14.1	100.0	48.0	100.0

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Material			Lead			Zinc			Silver			Remarks
Mesh	Weight, per cent	Cumulative weight, per cent	Assay, per cent	Distribution, per cent	Cumulative distribution, per cent	Assay, per cent	Distribution, per cent	Cumulative distribution, per cent	Assay, grams	Distribution, per cent	Cumulative distribution, per cent	
MILL HEADS												
On 35	2.5	2.5	1.4	0.29	0.29	10.5	2.21	2.21	73	0.64	0.64	-
On 48	5.6	8.1	2.2	1.14	1.43	11.8	5.61	7.82	75	1.48	2.12	-
On 65	10.2	18.3	3.1	3.03	4.47	12.7	11.05	18.87	100	3.50	5.72	-
On 100	12.3	30.6	4.5	5.23	9.70	11.3	11.82	30.69	178	7.73	13.45	-
On 150	10.8	41.4	9.0	9.22	18.92	13.3	12.24	42.93	246	9.39	22.84	-
On 200	10.6	52.0	11.6	11.69	30.61	13.5	12.15	55.09	303	11.35	34.19	-
Through 200	48.0	-	15.2	69.39	-	11.0	44.91	-	388	65.81	-	-
Total	100.0	100.0	10.5	100.00	100.00	11.8	100.00	100.00	283	100.00	100.00	-
MILL TAILING												
On 35	1.3	-	1.2	3.42	-	2.3	1.34	-	51	1.58	-	Magnetic.
Do	2.0	3.3	0.6	2.56	5.98	9.1	8.03	9.37	43	2.04	3.62	Nonmagnetic.
On 48	4.0	-	1.3	11.11	-	1.6	2.76	-	47	4.36	-	Magnetic.
Do	3.4	10.7	1.0	7.26	24.35	9.4	13.72	25.85	50	3.92	11.90	Nonmagnetic.
On 65	7.5	-	0.8	12.82	-	1.1	3.54	-	44	7.61	-	Magnetic.
Do	4.9	23.1	1.1	11.54	48.71	8.2	17.39	46.78	68	7.75	27.26	Nonmagnetic.
On 100	9.4	-	0.4	8.12	-	0.8	3.24	-	40	8.75	-	Magnetic.
Do	5.1	37.7	0.6	6.62	63.45	5.5	12.17	62.19	52	6.17	42.18	Nonmagnetic.
On 150	7.7	-	0.2	3.21	-	0.7	2.33	-	40	7.13	-	Magnetic.
Do	4.1	49.4	0.4	3.42	70.08	4.0	7.08	71.60	43	4.09	53.40	Nonmagnetic.
On 200	8.1	-	0.2	3.42	-	0.5	1.73	-	38	7.13	-	Magnetic.
Do	3.5	61.0	0.2	1.50	75.00	2.5	3.75	77.08	42	3.39	63.92	Nonmagnetic.
Through 200	18.1	-	0.3	11.54	-	0.4	3.11	-	42	17.66	-	Magnetic.
Do	20.9	-	0.3	13.46	-	2.2	19.81	-	38	18.42	-	Nonmagnetic.
Total	100.0	100.0	0.47	100.00	100.00	2.32	100.00	100.00	43	100.00	100.00	

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Table 4.- Screen-assay analyses of El Potosi concentrates

Material			Lead			Zinc			Silver			
Mesh	Weight, per cent	Cumulative weight, per cent	Assay, per cent	Distribution, per cent	Cumulative distribution per cent	Assay, per cent	Distribution, per cent	Cumulative distribution per cent	Assay, grams	Distribution, per cent	Cumulative distribution, per cent	Fe, P per cent
LEAD CONCENTRATE												
On 35	0.1	0.1	--	--	--	--	--	--	--	--	--	--
On 48	1.1	1.2	43.0	0.70	0.70	8.5	2.56	2.56	986	0.64	0.64	19.4
On 65	3.9	5.1	49.5	2.86	3.56	7.1	7.95	10.51	1267	2.93	3.57	16.2
On 100	7.9	13.0	59.2	6.94	10.50	5.5	12.22	22.73	1489	6.98	10.55	12.6
On 150	9.8	22.8	67.5	9.80	20.30	3.9	10.80	33.53	1656	9.63	20.18	8.5
On 200	11.2	34.0	71.3	11.85	32.15	3.2	10.23	43.76	1788	11.88	32.06	6.0
Through 200	66.0	--	69.3	67.85	--	3.0	56.24	--	1735	67.94	--	6.0
Total	100.0	100.0	67.4	100.00	100.00	3.5	100.00	100.00	1688	100.00	100.00	7.5

Examination of the coarser sizes of lead concentrate shows that grinding had not been sufficiently fine to unlock the minerals at these sizes. The iron mineral in lead concentrate is principally pyrrhotite.

ZINC CONCENTRATE												
On 35	0.6	0.6	0.6	0.70	0.70	46.0	0.59	0.59	40	0.55	0.56	16.1
On 48	3.4	4.0	0.8	4.72	5.42	47.1	3.39	3.98	42	3.35	3.91	15.7
On 65	9.0	13.0	0.9	14.16	19.58	47.1	8.99	12.97	44	9.29	13.20	15.7
On 100	13.0	26.0	0.7	15.91	35.49	47.1	12.98	25.95	45	13.73	26.93	15.7
On 150	12.4	38.4	0.6	12.94	48.43	47.1	12.39	38.34	46	13.37	40.30	15.8
On 200	13.4	51.8	0.4	9.44	57.87	47.5	13.51	51.85	46	14.45	54.75	15.7
Through 200	48.2	--	0.5	42.13	--	47.1	48.15	--	40	45.25	--	15.5
Total	100.0	100.0	0.57	100.00	100.00	47.15	100.00	100.00	42.6	100.00	100.00	15.6

It will be seen from the above screen analysis of zinc concentrates that the assay of the various screen sizes is about the same. This indicates that the zinc mineral is almost completely unlocked even at 35-mesh size. Of further interest is the analysis of iron in the various screen sizes, the highest being 16.1 per cent and the lowest 15.5 per cent, showing that the iron and zinc are chemically combined and that there is only a small amount of iron mineral other than marmatite in the concentrate.

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The following tables are given, showing the costs per ton for milling El Potosi ores for the several years operation:

Table 5.- Milling costs for previous years

	1925	1926	1927	1928	1929	1930	1931
Symons crushing	$0.056	$0.013	$0.019	$0.019	$0.021	$0.018	$0.014
Fine grinding	.202	.161	.125	.120	.088	.088	.074
Lead flotation reagents	.065	.060	.045	.035	.031	.043	.033
Zinc flotation reagents	.080	.126	.101	.098	.108	.084	.067
Flotation other expense	.104	.061	.057	.035	.032	.026	.026
Dewatering and loading Concentrate	.079	.047	.033	.033	.028	.026	.022
Power	.382	.324	.295	.271	.251	.254	.224
Water	.174	.116	.083	.076	.062	.052	.025
Assaying	.059	.051	.043	.039	.035	.029	.013
Miscellaneous other expense	.158	.041	.049	.035	.039	.028	.020
Tailings disposal	.015	.014	.009	.008	.014	.006	.008
General expense	.178	.157	.089	.093	.101	.091	.047
Total cost per ton	1.552	1.171	0.948	0.862	0.838	0.745	0.573
Total tons of ore milled	34,915	118,564	247,794	320,287	401,012	469,901	597,673
Material used per ton of ore:							
Rods, pounds	---	1.605	1.79	1.45	1.202	0.885	0.772
Cost cents	---	6.00	6.58	5.48	5.15	3.511	3.032
Liners, pounds	---	---	0.302	0.32	0.365	0.164	
Cost cents	---	---	3.20	3.83	3.54	1.60	
Power, kilowatt-hours per ton	---	22.90	22.00	20.43	18.333	14.626	13.903

Table 6.- Milling costs, 1932

	Six months, 1932	July, 1932				
		Labor	Supplies	Power	Reagents	Total
Symons crushing	$0.021	$0.0055	$0.0053	$0.0062	-	$0.0170
Fine grinding	.172	.0101	.0465	.1004	-	.1570
Flotation oils and reagents	.078	.0014	-	.0008	$0.0698	.0720
Flotation other expense	.117	.0119	.0119	.0934	-	.1172
Dewatering, storage, and loading	.030	.0146	.0072	.0078	-	.0290
Water	.020	.0034	.0012	.0125	-	.0171
Return water	.008	.0031	.0010	.0046	-	.0087
Tailings disposal	.010	.0044	.0045	.0018	-	.0107
Laboratory	.007	.0030	.0021	.0012	-	.0063
Research	.003	.0025	-	-	-	.0025
General expense	.074	.0519	.0080	.0059	-	.0658
Total	0.540	0.1118	0.0877	0.2346	0.0698	0.5039

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Table 7.- Costs in units of labor, power, and reagents[1/]

Labor

	Tons per 8-hour man-shift	Man-hour per ton
Shift foreman, 1 man per shift	783	0.0102
Crushing, 2 men per shift	391	.0205
Grinding and classification, 3 men per shift	261	.0307
Flotation, 2 men per shift	391	.0205
Sampling, 1 man per shift	783	.0102
Filtering, 4 men per shift	196	.0408
		0.1329
Miscellaneous. Includes superintendent, office, mechanics, warehouse, watchmen, pumpers, patio labor, assay office, school, etc.	33	0.2424
Total, all departments	21.3	0.3753

Power

	Kilowatt-hours per ton of ore milled	Per cent of total
Crushing and conveying to fine-ore bins	0.49	3.56
Grinding and classification	5.55	40.34
Flotation machines	4.61	33.50
Thickeners	.24	1.74
Wilfley sand pumps	.78	5.67
Filters	.32	2.33
Return-water pumps	.27	1.96
Fresh-water pumps	.75	5.45
Lighting	.37	2.69
Assay office	.14	1.02
Conveyors to rod mills and concentrate conveyors	.24	1.74
Total	13.76	100.00

Reagents

	Pound per ton ore milled
Cresylic acid	0.054
Ethyl xanthate	.125
Amyl (or butyl) xanthate	.110
Copper sulphate	.424
Sodium cyanide	.050
Lime	.479

1/ For rod and liner consumption, see Table 5.

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Table 8.- Metallurgical data for July, 1932

Dry ore treated	tons	53,285
Moisture in ore to mill	per cent	1.8
Hours operated per day		24
Days operated		23
Tons treated per 24 hours		2,317
Total concentrates produced	dry tons	18,543
Average concentrates produced per day:		
Lead	do	356
Zinc	do	459
Ratio of concentration of section:		
Lead	tons original ore into 1	6.51
Zinc	do	5.15
Average pulp density of heads:		
Lead section	percentage of solids	47.5
Zinc section	do	42.0
First zinc cleaner	do	40.1
Recleaner zinc section	do	46.6
Grinding-rod consumption per ton of ore	pound	0.542

Table 9.- Assays and recoveries of mill products

	Assays			Recoveries, per cent		
	Ag, ounces	Pb, per cent	Zn, per cent	Ag	Pb	Zn
Heads	8.44	11.35	11.40	100.00	100.00	100.00
Lead concentrates	48.12	68.20	3.39	87.96	95.65	4.57
Zinc concentrates	1.54	.73	46.44	3.55	1.25	79.18
Tailings	1.10	.54	2.84	8.49	3.10	16.25

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LEGEND OF EQUIPMENT

(1) 2,200-ton coarse-ore bin.
(2) 400-ton coarse-ore bin.
(3) Apron feeder.
(4) 24-inch belt conveyor.
(5) 24-inch belt conveyor.
(6) Merrick weightometer.
(7) Suspended magnet.
(8) 4 by 5 foot Hum-mer screen.
(9) 24-inch belt conveyor.
(10) Symons 5½-foot cone crusher.
(11) 24-inch belt conveyor.
(12) 18-inch belt conveyor.
(13) 600-ton fine-ore bin.
(14) 500-ton fine-ore bin.
(15) Drum-type ore feeder.
(16) Drum-type ore feeder.
(17) 12-inch fine-ore belt conveyor.
(18) Merrick weightometer.
(19) 14-inch fine-ore belt conveyor.
(20) Merrick weightometer.
(21) 12-inch belt conveyor.
(22) 2 Cole-Bergman 6 by 12 foot rod mills. Section 1.
(23) 2 Cole-Bergman 6 by 12 foot rod mills. Section 2.
(24) 2 Marcy 5 by 12 foot rod mills. Section 3.
(25) Dorr duplex classifiers, 6 by 22 foot.
(26) 14-cell, 18-inch, M.S. Sub-A flotation machine, for lead.
(27) 14-cell, 18-inch, M.S. Sub-A flotation machine, for lead.
(28) 14-cell, 18-inch, M.S. Sub-A flotation machine, for lead.
(29) 14-cell, 18-inch, M.S. Sub-A flotation machine, for lead.
(30) 12-cell, 24-inch, M.S. Standard Spitz flotation machine, zinc rougher.
(31) 12-cell, 24-inch, M.S. Standard Spitz flotation machine, first zinc cleaner. Divided to make two 6-cell machines.
(32) 12-cell, 24-inch, M.S. Standard Spitz flotation machine, second zinc cleaner. Divided to make two 6-cell machines.
(33) 12-cell, 24-inch, M.S. Standard Spitz flotation machine, zinc rougher.
(34) 12-cell, 24-inch, M.S. Standard Spitz flotation machine, zinc rougher.
(35) 12-cell, 24-inch, M.S. Standard Spitz flotation machine, zinc rougher.
(36) Four-way distributor for lead heads.
(37) Four-way distributor for zinc heads.
(38) Three-way distributor for cleaner tailings.
(39) Dorr Simplex classifier, 2 by 15 feet, for lead concentrate.
(40) Two center-drive Dorr thickeners, 30-foot diameter, for lead concentra
(41) Two 6-foot, four-leaf American filters, for lead concentrate.
(42) Dorr Simplex classifier, 3 by 15 feet, for zinc concentrate.
(43) Dorr traction thickener, 45 foot diameter, for zinc concentrate.
(44) 8 by 12 foot Dorrco filter, for zinc concentrate.
(45) 3 by 15 foot drag classifier, spare.
(46) Two Dorr center-drive thickeners, spares.
(47) Two 6-foot, four-leaf American filters, spares.

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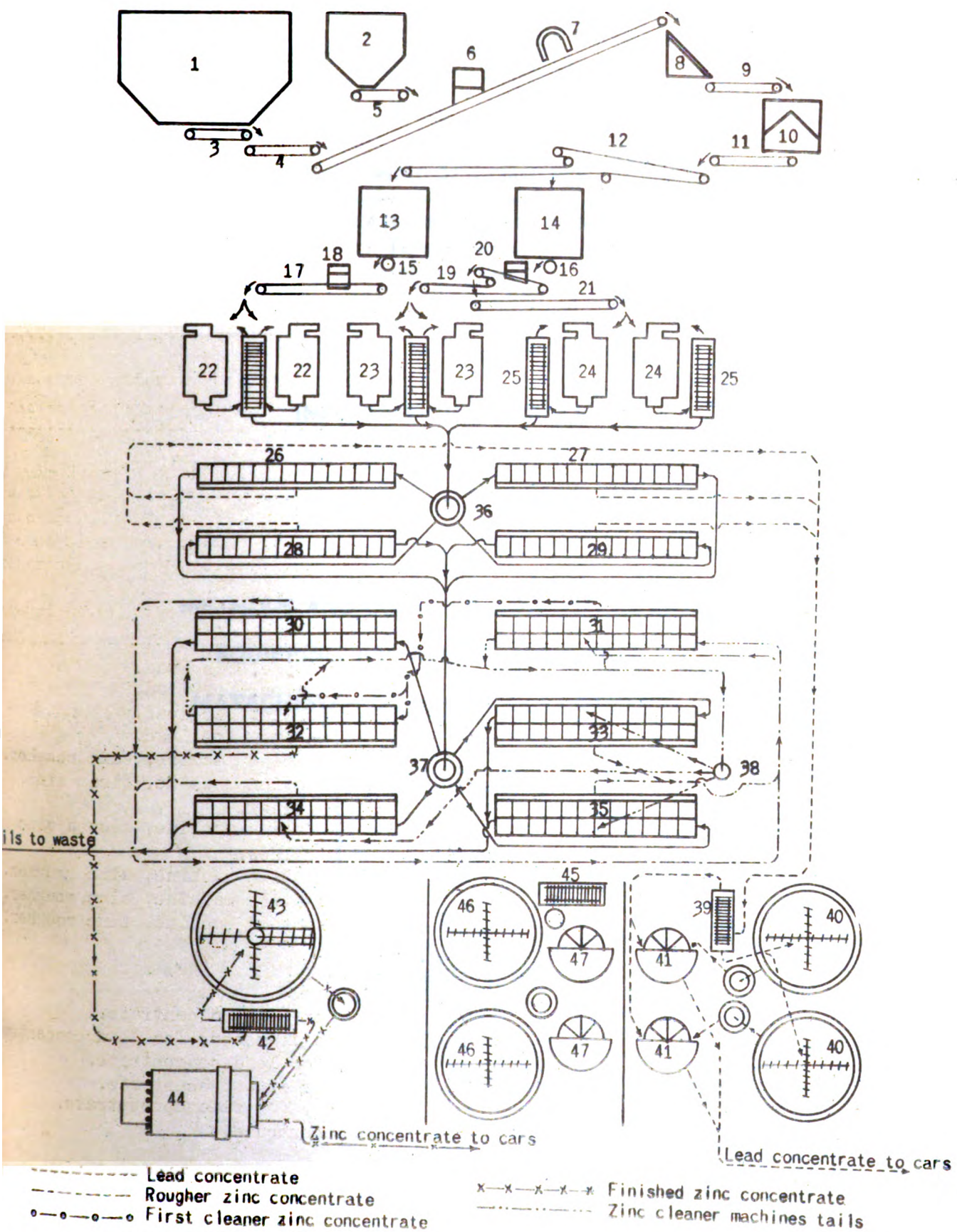

Figure 1.- Mill flow sheet

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